EMPIRICAL APPROACHES FOR WEAK ROCK MASSES

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ABSTRACT

A major focus of ground control research presently being conducted by the Spokane Research Laboratory of the National Institute for Occupational Safety and Health (NIOSH) is to incorporate weak rock masses (such as are associated with operations in the Carlin Trend in Nevada) into existing design relationships. The original database that led to most of the empirical design relationships presently employed in hard-rock mining was derived from fair-to-good-quality rock. In this study, the relationship between weak rock quality and opening design (nonentry/entry methods) is being investigated. The common factor in all mines is a weak back or wall. This work attempts to provide tools that will enable a mine operator to make economic decisions that will also ensure a safe working environment.

INTRODUCTION

Researchers at the Spokane Research Laboratory of the National Institute for Occupational Safety and Health, Spokane, WA, and the University of British Columbia, Vancouver, B.C., have assembled a team to develop underground design guidelines for safe and cost-effective mining within a weak rock mass. Such work also includes developing novel support methods, such as the use of synthetic-fiber-reinforced shotcrete, ways to undermine underhand-and-fill backfilled stopes, and assess supports presently in place in weak rock masses. In the present study, rock mass interaction with grouted bolt supports was investigated in three mines in Nevada and backfill, pillar, and bolt support were studied in one.

Many Nevada gold deposits are found in intensely fractured, faulted, and argillized host rock. As a result, underground mining is often difficult and hazardous, as indicated by the number of injuries and fatalities from uncontrolled falls of ground (Table 1).

Table 1.—Ground control injuries and fatalities in underground gold mines in Nevada, 1985 through 2000.

Fatalities	7
Permanent disabilities	4
Lost-time injuries	49
Restricted-activity injuries	46
Other injuries	110
Total	216
Reported rock falls with no injuries[1]	69

[1] Includes MSHA data for noninjury incidents where a reportable fall of ground occurred but did not cause injuries because the mining area was unoccupied.

A comparative analysis by the Mine Safety and Health Administration (MSHA) for the years 1990 through 1999 indicated that the number of roof fall injuries in Nevada has varied from a low of eight in 1991 to a high of 28 in 1995 and 1997. As late as 1999, the number of injuries was still in the double digits (Figure 1).

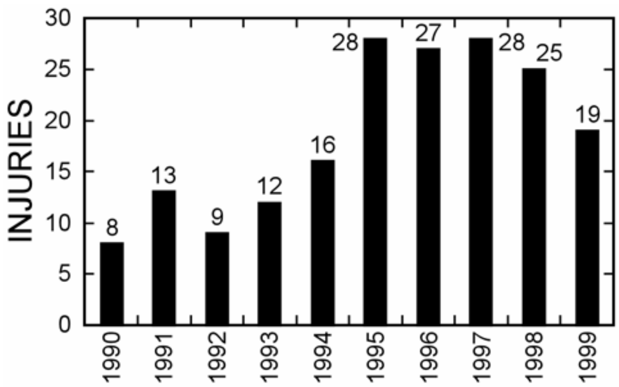


Figure 1.—Injuries in Nevada, 1990-1999 (Hoch 2000).

Mining is a dynamic process, and ground conditions can change over a short distance. A mine opening must perform in a predictable manner over its expected life. Empirical design methods have been used over the past 30 years because they permit the overall behavior of a rock mass to be predicted easily and accurately. The basis for the success of the empirical method is a strong foundation of field data coupled with ongoing field observations that allow changing rock conditions to be evaluated as mining progresses. Based on the collection and analysis of case histories, the relative influence of factors such as joint orientation, joint strength, and joint spacing on stability can be estimated. The original database of case histories exhibited a Rock Mass Rating (Bieniawski 1976) (RMR_{76}) in excess of 50% or Q values of 2.0 or greater.

Any method of designing an opening must be easy to assess, understand, applied, modified if necessary, and reproduced for the next application. A critical factor is that the design incorporate the degree of stability required of any mine entry.

SPAN DESIGN MAN ENTRY METHODS

The "critical span curve" was a concept developed in 1994 to evaluate back stability in cut-and-fill mines (Lang 1994). In 2000, the span curve database developed by the University of British Columbia was expanded to include 292 case histories from mines primarily in Canada (Wang et al. 2000). The information from these case histories provides the basis for the span design curve shown in Figure 2.

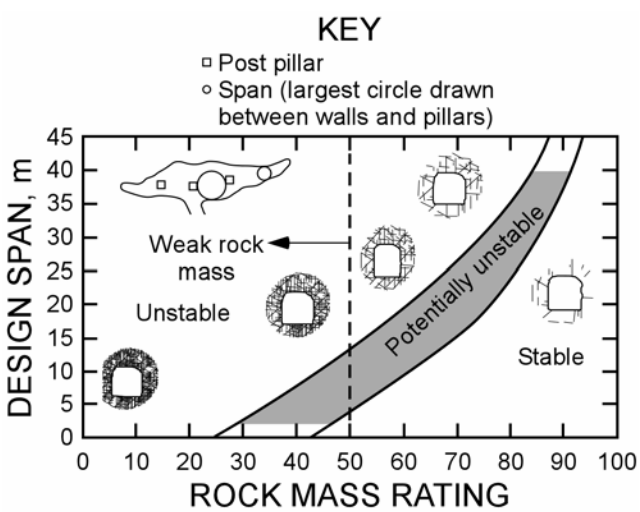


Figure 2—Design span curve.

A "critical span" is defined as the diameter of the largest circle that can be drawn within the boundaries of the exposed back. The stability of this exposed span is related to the type of rock in the immediate back. The "design span" refers to backs that have no support and/or spans that are supported with localized pattern bolting (1.8-m-long bolts on 1.2- by 1.2-m spacings). Excavation stability is classified into three categories:

1. Stable excavation.
   a. No uncontrolled falls of ground.
   b. No movement of the back is observed.

   c. No extraordinary support measures have been employed.
2. Potentially unstable excavation.
   a. Extra ground support has been installed to prevent potential falls of ground.
   b. Movement has occurred in the back.
   c. Increased frequency of ground movement has been observed.
3. Unstable excavation.
   a. Area has collapsed.
   b. Depth of failure of the back is 0.5 times the span (in the absence of major structures).
   c. Support was not effective in maintaining stability.

A minus-10 correction factor is applied to the final RMR_{76} value when evaluating rock with shallow-dipping or flat joints. However, the applicability of this factor is being reassessed for weak ground because of its amorphous nature and because joint direction is expected to play a minor role. Where discrete ground wedges have been observed and identified, they must be supported prior to employing the critical span curve.

Stability is generally defined in terms of short-term stability because the database is based largely on stoping methods that, by their nature, are of short duration. Movement of the back greater than 1 mm within a 24-hour period has also been defined as a critical amount of movement for safe access (Pakalnis 2002). In Figure 3*A*, RMR values are plotted against number of occurrences. Less than 10% fall below RMR values of 40%, and less than 20% fall below values of 55%.

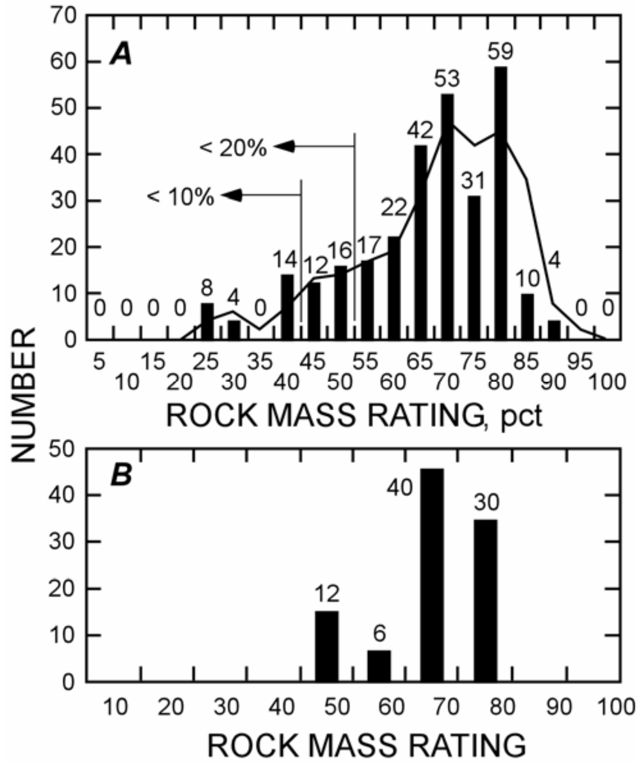
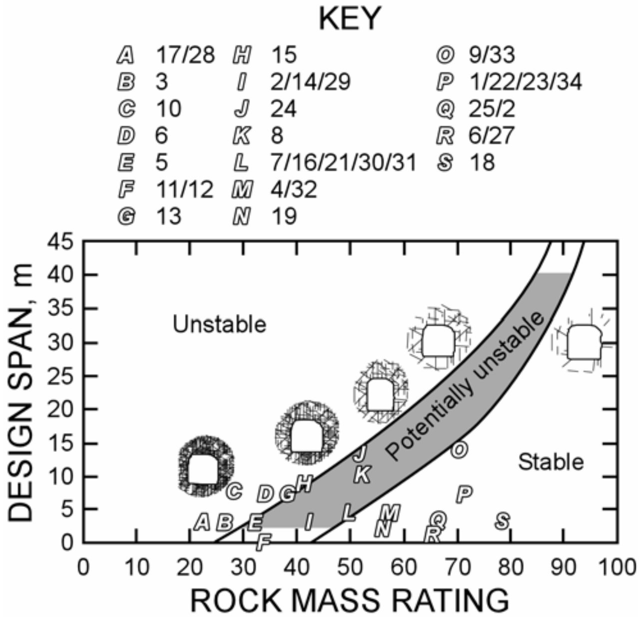


Figure 3.—Distribution of RMR. *A*, Span database (Lang 1994); *B*, stability graph database (ELOS) (Clark and Pakalnis 1997).

Some 36 case histories from five different mines with RMR values varying between 20 and 85 were added to the information base for the critical span curve (Table 2). Several values were less than 55% RMR; the lowest RMR value calculated for any location was 25% (Figure 4). This information was used to augment the original "Span design curve for man-entry" (Lang 1994) mining as shown in Figure 4. The span curve enables an operator to assess back stability with respect to a rock mass. The information has been used successfully to predict the stability of weak rock masses and has provided operators with an additional design tool for making decisions about mine openings. The data are being coupled with depth of failure to define the amount of support required to arrive at a safe, cost-effective man-entry design.

Table 2.—Augmented span database from NIOSH

Location	RMR	Q rating	Back span, m	Support
Mine 1:				
1	70	10.00	4.6	2.4-m rebar, 1.8- × 1.5-m pattern, SS39 2.4 m long, 0.9- × 0.9- pattern, no shotcrete
2	40	2.10	4.6	Stable with shotcrete plus 2.4-m rebar and 2.4-m SS39
3	25	0.06	4.6	Stable with shotcrete plus 4.9-m cables and Swellex bolts SS39 2.4-m long, 0.9- × 0.9-m pattern. Permanent opening.
4	55	4.20	5.5	2.4-m rebar, 1.8- × 1.5-m pattern. 2.4-m-long Swellex bolts, 1.5- × 0.6-m pattern. No. 6 gauge weldmesh plus 5.2-cm shotcrete.
5	30	0.75	6.1	Stable with shotcrete plus 2.4-m rebar and 2.4-m SS39.
6	30	0.75	6.1	Stable with shotcrete plus 2.4-m rebar and 2.4-m SS39.
7	45	2.10	4.6	2.4-m rebar, 1.8- × 1.5-m pattern, SS39 2.4-m long, 0.9- × 0.9-m pattern. No shotcrete.
8	50	2.50	6.1	Bolts and mesh plus shotcrete.
9	70	35.00	11.3	Cable plus shotcrete. Very stable.
10	25	0.30	7.3	Stable with shotcrete plus 3.7-m rebar or cable bolt or SuperSwellex bolts on 2.4- × 2.4-m pattern.
11	30	0.20	3.0	Caved, unsupported to face over 6.1-m span.
12	30	0.20	1.8	Stable, unsupported to face over 6.1-m span.
Mine 2:				
13	20	0.7	3.3	Caved
14	85	95.10	3.3	Stable
15	55	3.40	3.0	Stable
16	60	5.90	12.2	Caved, ~6 m in back
17	50	5.90	3.7	Potentially unstable. Stable with support.
18	74	28.00	3.7	Stable
19	70	18.00	3.7	Stable
20	50	1.90	12.0	Stable with cable bolts
21	64	9.20	3.7	Stable
22	65	10.30	3.7	Stable
23	65	10.30	3.7	Stable
24	43	0.89	4.6	Caved to 2.4 m above back.
Mine 3:				
25	40	0.64	6.1	Caved with support
26	40	0.64	4.9	Stable with 1.5-m SS39 on 0.9- × 0.9-m pattern plus 1.5-cm shotcrete
Mine 4:				
27	40 (30-52)	0.64	4.6	Stable with support
28	55	3.40	7.6	Stable with support
Mine 5				
29	25	0.10	2.4	Caved, no support.
30	25	0.10	2.4	Stable with 1.5-m SS39 on 0.9- × 0.9-m pattern plus 1.5-cm shotcrete
31	45	1.10	3.8	Potentially unstable. 2.4-m bolts on 0.9- × 0.9-m pattern plus 10.2-cm-wide straps.
32	50	1.90	3.8	Stable
33	55	3.40	3.8	Stable
34	60	5.90	5.0	Stable
35	75	31.30	5.0	Stable
36	75	31.30	20.0	Stable



[1]Figure 4.—Augmented span curve. Numbers in key correspond to mine numbers in Table 2. Letters indicate location on the span curve.

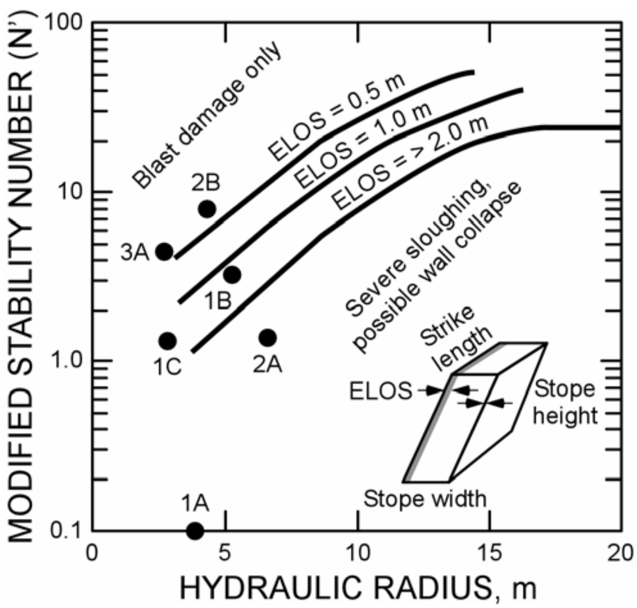


Figure 5.—Stability graph.

STABILITY GRAPH METHOD, NON-ENTRY MINING

The original stability method for open stope design was based largely on Canadian operations and proposed by Matthews et al. in 1981, modified by Potvin (1988), and updated by Nickson (1992). In all instances, stability was qualitatively assessed as either being stable, potentially unstable, or caved. Recent research by Mah (1997) and Clark and Pakalnis (1997) at the University of British Columbia has augmented the stability graph by using stope surveys in which cavity monitoring systems were employed. This research has enabled the determination of the amount of dilution. A parameter defined by Clark (1998) as the equivalent linear overbreak/slough (ELOS) (Figure 5) was used to express volumetric measurements of overbreak as an average depth over the entire stope surface. ELOS is defined as the volume of slough from the stope surface divided by the product of stope height times wall strike length.

The stability graph relates hydraulic radius (HR) of the stope wall to empirical estimates of overbreak slough. The graph was originally based on 88 observations from Clark.(1998). The distribution of rock quality is summarized in Figure 3*B*, which shows the limited amount of data for lower rock mass values. HR is defined as the surface area of an opening divided by perimeter of the exposed wall being analyzed.

Table 3 summarizes the data from six locations at the three mines having lower RMR values. The values from Table 3 are plotted in Figure 5. The infinity symbol (mine 2A) means the stope wall is undercut and is therefore defined as having an infinite height. The following equation was employed for calculation of parameters for the database shown in Figure 5.

$$N' = Q' * A * B * C$$

where N' = Modified stability number,

Q’ = NGI rock quality index (after Barton et al. 1974),
A* = High stress reduction,
B* = Orientation of discontinuities,
and C* = Orientation of surface.

Table 3.—Augmented stability graph for stope walls from NIOSH

Mine	Rock type	N[1]	RMR	Q	Height-width, m	B	Dip, deg	Hydraulic radius	Comments
1A	Limestone	0.1	25	0.06	18 by 18	0.3	90	4.6	Cave
1B	Limestone	1.8	30	0.75	18 by 23	0.3	90	5.1	Caved
1C	Limestone	1.8	30	0.75	12 by 23	0.3	90	4.0	Stabl;e
2A	Mafics	1.5	50	1.95	12 by ∞[2]	0.2	50	6.1	Caved/undercut
2B	Mafics	8.4	65	10.30	15 by 22	0.2	50	4.5	Stable
3A	Limestone	5.1	40	0.64	14 by 20	1.0	50	4.1	Stable

[1] A = 1 (relaxed height-width). [2] Undercut height-width.

The stress reduction factor and joint water reduction factor are equal to 1, as they are accounted for separately within the analysis.

For the ELOS graph points, the database for the stability graph was derived from mining operations that are generally dry.

The following relationship was used to convert RMR to Q’ (from Bieniawski 1979):

$$RMR = 9 \mathrm{Ln} Q' + 44.$$

The A factor accounts for the influence of high stresses that reduce rock mass stability and is determined by the ratio of unconfined compressive strength of intact rock to maximum induced stress parallel to the opening surface. It is set to 1.0 if intact rock strength is 10 or more times induced stress, which indicates that high stress is not a problem. It is set to 0.1 if rock strength is two times induced stress or less, which indicates that high stress significantly reduces opening stability. In the mines visited for this study, the value of A was equal to 1.0 because the hanging wall was largely in a relaxed state.

The B factor looks at the influence of the orientation of discontinuities with respect to the surface analyzed and states that joints oriented 90° to a surface do not create stability problems. The B factor is set to 1.0. Discontinuities dipping up to 20° to the surface are the least stable and represent geologic structures that can topple. In this case, B is equal to 0.2, which was the value used for the database. In extremely weak rock masses (RMR = 25%), the material largely resembles an amorphous mass with geologic structures throughout, therefore reduction due to jointing is suspect. The authors are presently analyzing the data to augment this factor.

The C factor considers orientation of the surface being analyzed and is assigned a value of 8 for the design of vertical walls and a value of 2 for horizontal backs. The C factor reflects the inherently more stable nature of vertical walls compared to a horizontal back. In this paper, the ELOS curves employ a value of C = 8.0 for the footwalls. For a more complete explanation of factors A, B, and C, refer to papers by Potvin (1988) and Clark and Pakalins (1997).

SUPPORT GUIDELINES

The development of support guidelines are critical. Both Table 1 and Figure 1, which are based on MSHA data, show that the majority of mine fatalities and serious injuries are a result of rock falls from the back or immediate roof. Ground support in weak rock presents special challenges. Under-design can lead to costly failures, whereas over-design can lead to high ground control support costs.

Figure 6 depicts a classic wedge failure if the ground support has been under-designed using bolts.

Over 400,000 Split-Set[1] friction bolts are used in Nevada mines as primary support (Goris 2003). Friction bolts are particularly useful in fissile, buckling, or sheared ground where it is difficult to secure a point anchor. An analysis of the performance of friction bolts in mines with weak rock (as determined by the RMR index) needed to be addressed. Most Nevada mines use 39-mm Split-Set bolts, whereas mines in Canada use 33-mm Split-Set bolts. Also, mines in Canada generally use these bolts only in the walls and not in the back.

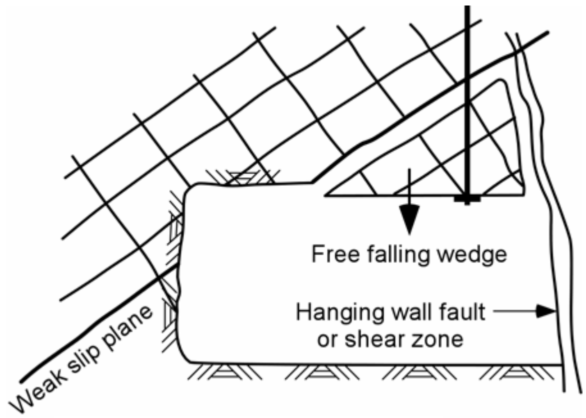


Figure 6.—Classic wedge failures

Table 4 shows updated support property values and includes data from the 39-mm Split-Set friction bolts.

A neural net (Ward Systems 2003) was superimposed on the mine data to determine if trends or predictions could be made with unrelated field data. The neural net methodology has been used in establishing design curves for span and stope design by Wang et al. (2002)

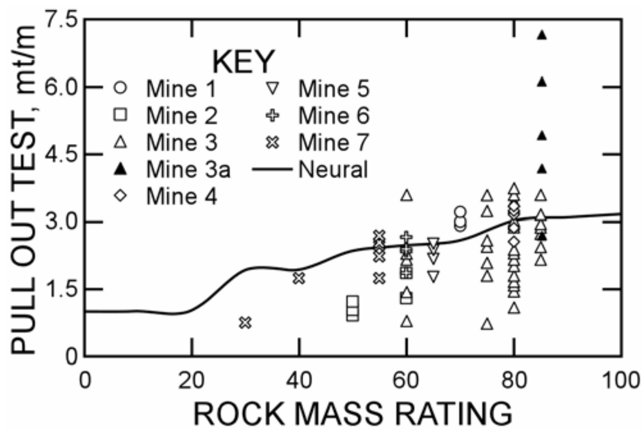


Figure 7.—RMR versus pull-out strength. A neural trend line is superimposed.

Data points gathered from several pull tests in weak rock masses were plotted as shown in Figure 7. The graph shows a strong trend between RMR and bond strength; this relationship is being assessed as part of the ongoing research. Variability in the test results shows the difficulty in assessing overall support for a given heading. Thus, it is imperative that mines develop a database about the support used in mines so they can analyze unexpected ground conditions.

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[1] Mention of specific products and manufacturers does not imply endorsement by the National Institute for Occupational Safety and Health.

Table 4.—Support properties

Rock bolt:[1]	Yield strength, mt	Breaking strength, mt
Mechanical, 1.59 cm	6.1	10.2
Split-Set (SS 33)	8.5	10.6
Split-Set (SS39)	12.7	14.0
Standard Swellex	NA	11.0
Yielding Swellex	NA	9.50
Super Swellex	NA	22.0
Rebar, 20 mm, No. 6	12.4	18.5
Rebar, 22 mm, No. 7	16.0	23.0
Rebar, 25 mm, No. 8	20.5	30.8
Dywidag, No. 6	11.9	18.0
Dywidag, No. 7	16.3	24.5
Dywidag, No. 8	21.5	32.3
Dywidag, No. 9	27.5	40.9
Dywidag, No. 10	34.6	52.0
Cable bolt, 1.27 cm	15.9	18.8
Cable bolt, 1.59 cm	21.6	25.5
Screen:[2]	**Bag strength, mt**[3]	
Welded wire mesh, 4 gauge	3.6	
Welded wire mesh, 6 gauge	3.3	
Welded wire mesh, 9 gauge	1.9	
Welded wire mesh, 12 gauge	1.4	
Chainlink, 11 gauge, bare metal	2.9	
Chainlink, 11 gauge, galvanized	1.7	
Chainlink, 9 gauge, bare metal	3.7	
Chainlink, 9 gauge, galvanized	3.2	
Bond strength, mt/m		
Split-Set, hard ground	2.4-4.9	
Split-Set, weak ground	0.8-3.7	
Swellex, hard ground	3.9-14.0	
Cable bolt, 1.59 cm, hard ground	26-∞0.91 m	
Rebar, No. 6, hard ground	18-∞30.5 cm, granite	
Split-Set SS39, pull-out test in concrete	4.8	

[1]No. 6 = 19 mm; No. 7 = 22 mm; No. 8 = 25 mm; No. 9 = 29 mm; No. 10 = 32 mm.

[2]4 gauge = 5.1 mm diam; 6 gauge = 5.08 mm diam; 9 gauge = 4.1 mm diam; 11 gauge = 3.2 mm diam; 12 = 2.8 mm diam.

[3]Bag strength on 1.2- by 1.2-m pattern. Shotcrete shear strength = 2 MPa = 200 mt/m^2

CONCLUSIONS

Weak ground conditions in several mines were observed, and RMR values were calculated to update both span design calculations and stability graphs. The immediate rock mass was characterized and analyzed in terms of prevailing type of ground support, potential failure mechanisms, and rock behavior. An approach was employed that would permit an operator to understand overall failure mechanisms and resultant loads that would affect the system. This approach would allow an engineer to develop an optimum support strategy.

Variability in field conditions (rock mass, stress, geological structures, different types of ground support) showed the difficulty in assessing overall support for a given heading. It is imperative that mines develop a database on the support used in their mines so unexpected ground conditions could be analyzed. The results from these augmented design curves and pull-out tests are presented in hopes that they will aid mine professionals in their task of designing a safe work place.

Future work will include upgrading the relationship between RMR and support performance and collecting additional data for mines with RMR values below 50% to enable the calculation of more accurate stability graphs and span design curves. The overall objective of this research is the development of a safe and cost-effective work environment.

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